MINING HEALTH AND SAFETY UPDATE

NIOSH

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Improving Surface Mine Truck Haulage Safety

Powered haulage has been a leading cause of injuries and fatalities in surface mines, both coal and metal/nonmetal.

From 1989 to 1991, the greatest number of injuries and fatalities in U.S. surface mines involved haulage trucks. One of the most frequently cited factors contributing to the 100 fatalities recorded for 1995 was powered haulage. In the first quarter of 1997, 22 fatalities occurred; powered haulage contributed to half of them.

A research project at the NIOSH Spokane Research Laboratory directly addresses truck

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Predicting Methane Emissions on Extended Longwall Faces

The explosion or ignition of methane has been a major cause of multiple fatalities in U.S. underground coal mines. Several factors are known to influence methane gas emissions during longwall mining, including (1) coal extraction rate, (2) gas content of the coal mined, (3) methane drainage practices, (4) the proximity of surrounding gas-bearing strata to the mined coalbed, and (5) the development of mining-induced fractures that provide pathways for

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This issue's focus:

HAULAGE
DISASTER PREVENTION
GROUND CONTROL

U.S. DEPARTMENT OF HEALTH AND HUMAN SERVICES

Public Health Service
Centers for Disease Control and Prevention
National Institute for Occupational Safety and Health

CDC
CENTERS FOR DISEASE CONTROL AND PREVENTION

Improving Surface Mine Truck Haulage Safety

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haulage injuries and fatalities in surface mines. Studies have initially focused on technologies that would improve a haulage truck driver's field of vision while backing up. Several technologies investigated by contractors for the former U.S. Bureau of Mines (USBM) in the late 1970's and early 1980's were reexamined to determine their current status and whether they could be updated with current technology. In addition, one new technology, closed-circuit television (CCTV), was reviewed.

Video Camera Backup Monitor

A black-and-white CCTV system was recently installed on a 190-ton haulage truck and tested at an operating mine. The rear-mounted camera successfully withstood a year's cycle of loading, hauling, and dumping, as well as the rigors of desert heat and cold. The mine's truck shop fabricated a floor-mounted frame to hold a 6- by 7.75-in monitor, as well as a two-way radio and a truck-monitoring terminal. The camera is mounted on the rear axle of the frame, and the camera monitor is wired into the dash light system to reduce monitor glare during night shifts. The screen is black until the truck is shifted into reverse. There is also a manual mode that leaves the monitor on continually.

Since the initial test, the mine has installed cameras on all of its 190-ton trucks and is testing the use of a second camera on the right front deck railing to capture the right-hand blind side view. The monitor can be toggled to switch views from the right-hand to the rear-facing cameras. A split-screen monitor, showing both views at once, is being considered.

When implementing new technology such as cameras, mine personnel have stressed the importance of working with a supplier or manufacturer that may already be servicing other electronic components (e.g., radios) at the mine. In-mine startup and servicing support were cited as crucial to the successful use and modification of the technology to the mine's specific needs.

Blind Area Viewer

The blind area viewer was developed under a USBM contract. This device consisted of a finely etched Fresnel lens sandwiched between safety glass and housed in a protective, glare-resistant, louvered box. The Fresnel lens enabled a 70° downward and 60° horizontal view for the truck driver. When mounted on the right front deck of a large haulage truck, the viewer showed the blind area on the right side to within 5 ft of the truck. The blind area viewer was slightly modified after field trials. The lack of moving parts and limited need for power or electronic support made the viewer an attractive instrument for mines and was considered successful enough to go into commercial production. Currently, however, the blind area viewer is no longer being manufactured or available for distribution.

Frame housing monitor, radio, and terminal installed on a haulage truck.

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Doppler Radar Backup Alarm

A series of laboratory tests, controlled simulated tests on small vehicles, and full-scale field tests on 240-ton class, end-dump haulage trucks were conducted to evaluate Doppler radar discriminating alarms. Although these devices show much promise for sensing objects within the blind spots of vehicles, they are not just bolt-on and plug-and-play units that are easy to adapt to various situations. During the tests, it was shown that mounting the units to achieve optimum sensing ability without interference and false alarms from other systems was, at the very least, a tedious and time-consuming job. Although a magnetic mounting bracket helps determine the most efficient mounting location, the actual work of gain adjusting, aiming and pointing, evaluating the height for the sensor, and factoring out vibration and interference must be established through on-truck trial and error. In addition, how the devices performed when the truck was stationary did not indicate how well the devices worked when the truck or objects behind the truck were moving.

Summary

Powered haulage has been and continues to be a significant factor in injuries and fatalities in U.S. surface mines. Off-road haulage trucks continue to increase in size and capacity, and a 320-ton-capacity truck is now commercially available. Advances in electronics and other monitoring systems warrant the reexamination of promising technologies that could increase safety around these large trucks. At the same time, it must be noted that whiz-bang electronics, on-board computer terminals, and heat-treated titanium alloys do not make a safe mine. Safety equipment is just an added tool to an overall safety program. Use of new and different instruments is secondary to the practice of a sound mine safety plan.

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Installing dual radar units on truck chassis.

Predicting Methane Emissions on Extended Longwall Faces

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gas migration to the active face or gob areas. One factor for which the methane emission consequences were largely unknown was the industry's trend of mining increasingly larger longwall panels in an effort to improve mining efficiency. Unfortunately, gains in mining efficiency, along with the extraction of deeper, gassier coalbeds, may be accompanied by an increased flow of methane into the underground workplace.

The ability to predict the methane emission consequences of altering mine design parameters, such as longwall face width, is desirable for maintaining a safe workplace environment. If higher emissions rates are the result of changing mining practices, it is preferable from a safety perspective to be prepared in advance, with either increased ventilation airflow or appropriate methane drainage capabilities.

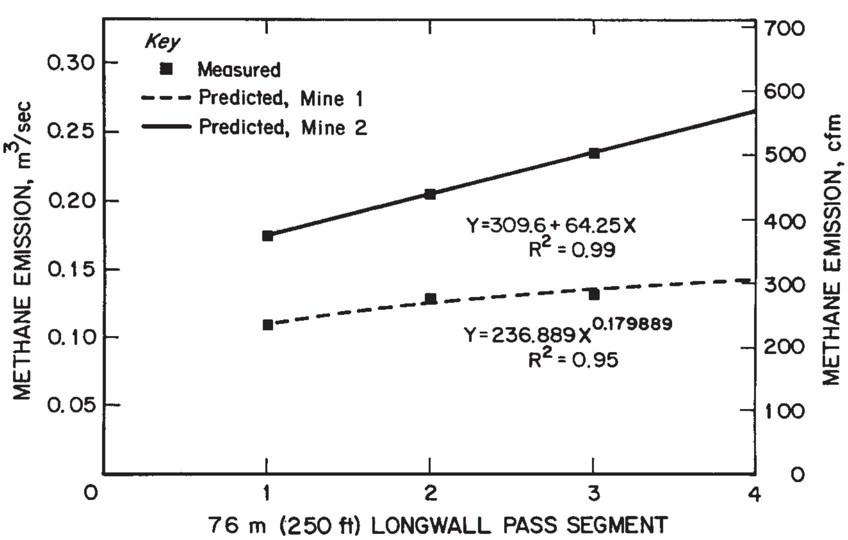
Researchers from the NIOSH Pittsburgh Research Laboratory conducted methane emission characterization studies at two adjacent mines operating in the Pocahontas No. 3 Coalbed in Virginia to study the emission consequence of extending longwall faces to greater width. Longwall face widths at these mines were to be increased from 750 to 1,000 ft. However, because historically high methane emissions from the longwall face and gob were already being experienced, there was a concern for further increases in emission rates.

To predict the methane emission consequences of mining 1,000-ft faces at the study mines, it was first necessary to measure current emission rates to determine if a general increase in methane emission levels occurs progressively during the mining of a longwall pass. Characterizing methane emission trends across the longwall face was facilitated by dividing each pass on the 750-ft-wide faces into three equal-length segments of 250 ft and determining the average face emission rate during the mining of each pass segment.

Predictions of methane emission levels for faces extended to 1,000 ft were made by regression analysis of the average longwall pass segment emission values from the 750-ft-wide faces at each mine. Projecting the regression curve to a fourth 250-ft longwall pass segment at Mine 1 yields an average methane emission rate of 304 cfm, or a 7% (20-cfm) increase for a 1,000-ft-wide face. At Mine 2, projecting the regression curve to a fourth 250-ft longwall pass segment yields an average methane emission rate of 567 cfm, or a 13% (65-cfm) increase for a 1,000-ft-wide face.

The relatively low predicted increase in methane emissions at the face for the first mine site would probably not result in additional methane control problems. However, it should be noted that the predicted average emission value is not necessarily the "worst-case" scenario, which would include high methane emission events such as the floor gas bleeders common at this mine. In contrast, the predicted average face emissions for 1,000-ft-wide faces at the second mine site (567 cfm) are within the general range where methane-related mining delays were experienced during the study.

Face widths of 1,000 ft have not been developed at the first mine site; therefore, the accuracy of the prediction could not be tested. Larger panels with 1,000-ft-wide faces were developed at the second mine site without experiencing the predicted higher methane emission levels. The mine's engineering staff credits an enhanced methane drainage program that provided both increased hole length and additional time for horizontal boreholes to drain gas in advance of mining with maintaining methane emissions at a lower than predicted level.



Methane emission prediction curves for 1,000-ft longwall faces in two mines operating in the Pocahontas No. 3 Coalbed in Virginia.

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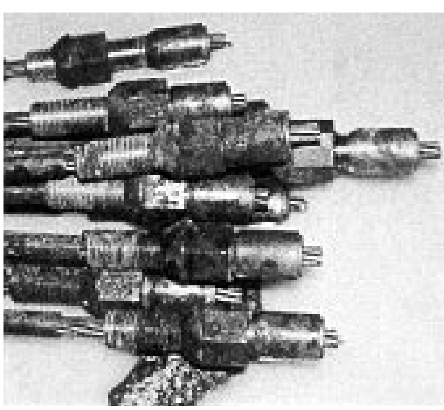
Safe Installation of Cable Bolt Supports Using Pneumatic Jacklegs

Cable bolts have been used for ground support in the mining industry for more than 50 years. The first cables were old wire ropes from shaft hoists that were cut to length and then degreased so that they could be used with cement-based grouts. During the 1970's, high-strength prestressing cable adopted from the building of concrete bridges was introduced. Having the capability of selecting the physical properties of the cables according to anticipated use provided a new means of secondary ground control. Also, because the cables were flexible, they could be inserted up to 20 m into a rock mass from a drift only 2 m in diameter. Thus, they could be used to replace timbers, section bolts, and low-yield bolts in low, narrow stopes and haulageways. Since the early 1990's, the use of resin-grouted cable bolts has increased significantly.

With all of the ground control combinations being used in coal and nonmetal mines, a cable bolt system that could be used in underground hard-rock mines to support the back and/or pillar was a logical step. In the coal and soda ash mining industries, 5- to 7-m-long bolts are inserted in the back by bolters. In hard-rock mines, however, the lack of space caused the emphasis to shift from placing bolts using bolters alone to using pneumatic jacklegs. Researchers have now developed a modified cable bolt support that incorporates a specialized inserter-tensioner head that allows the cable to be installed easily with a jackleg and then tensioned, if necessary. The bolt consists of a 15-mm-diam, 3-m-long cable fitted with a special threaded barrel-and-wedge head, nut, and support collar. It also incorporates Garford bulbs (i.e., bulbs or nodes spaced along the cable) to increase pullout resistance and enhance the mixing of resin. In addition, the cable has a stiffener to prevent cable whip during installation.

Researchers from the NIOSH Spokane Research Laboratory conducted a series of tests to study the safety of various insertion and handling techniques, as well as the pullout strengths of jackleg-installed cable bolts. Cables were tested in drill holes with diameters of 25, 32, and 42 mm and resin embedment lengths from 90 to 183 cm.

The tests proved the ground support system to be effective when the proper resin embedment length, hole diameter, and resin cartridge diameter were tailored for specific mines and test blocks. The tests also provided enough data to indicate that using a jackleg to insert cable bolts into resin cartridges is a reliable method of installation. However, the tests also showed that all cables to be installed in a mine should undergo a vigorous testing schedule geared to that specific mine to determine proper resin diameter and length before they are used. Some conditions pinpointed as likely to cause problems were rock fragmentation, the presence of bedding planes, and wet and squeezing holes.



Cable bolts with inserter-tensioner head and support collars for jackleg installation.

On the basis of problems experienced while conducting these tests, a set of safety guidelines is proposed below. These procedures should be followed to reduce the potential for injury while installing cable bolts with a jackleg.

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Safe Installation Procedures for Cable Bolts

Before Drilling

- Always wear safety boots, glasses, hat, and gloves.
- Cables are shipped 10 to a bundle and are very heavy, so take care to avoid back injuries when lifting.
- Resin cartridges are shipped in heavy boxes, so lift the resin boxes with care also.
- Be properly trained in the use of jackleg drills, as they are heavy and awkward to move to a drilling location. Avoid back injuries by using proper lifting techniques.
- Check airline fittings and hoses for proper fitting and cuts. Always use airline safety leashes.
- Check jackleg drill chuck and drill steel puller for burrs and file them off as needed.
- Make sure the foot of the jackleg is seated well before collaring, drilling, or inserting a cable bolt. A stuck drill will cause the leg to spin out of the floor, which could result in injury to the driller or someone standing nearby.
- Cover the drill hole with a glove or rag during blowout to protect eyes from flying debris.

During Installation

- While installing resin, take care not to puncture the resin tubes. Resin causes skin burns, and broken resin tubes do not provide proper bolt anchorage.
- Carefully install the cable bolt into the drill hole until resistance of the resin is felt, then place the other end into the jackleg fitted with a driving socket.
- Drive the cable into the hole first by spinning the cable, then adding air to the pusher leg. Slowly screw the leg in. Caution should be taken if the cable stops, because spring energy in the cable will cause the jackleg to bind.
- After the cable has seated against the plate and back or rib, spin the resin an additional 15 seconds and hold the bolt to the back or rib 30 more seconds while the toe resin sets up.

Factors That Influence Coal Mine Bumps

Coal bumps are violent failures of highly stressed coal. These can lead to serious injuries to underground mine workers. In an attempt to identify the most significant factors contributing to coal bumps, researchers from private industry and the NIOSH Spokane Research Laboratory performed statistical analyses of in-mine monitoring data from 25 sites in 6 room-and-pillar coal mines and 19 longwall mines in Colorado, Utah, Virginia, and Kentucky.

Engineers and researchers use two methods to help predict future bump conditions: computational and statistical. In this study, the team combined the strength of both methods to identify important variables and develop predictive capabilities. Computational methods were used to assess the influence of a combination of geometric variables into single variables, such as pillar factor of safety and released energy. This was very useful for increasing goodness of fit and enhancing multiple regression coefficients. Statistical methods, on the other hand, are uniquely capable of being applied where there are good data, but a limited understanding of certain natural phenomena, such as coal bumps. Statistical methods were used to identify significant variables, build confidence intervals, etc.

Identifying Potential Factors and Determining Bivariate Correlation Coefficients

The first step involved the identification of 25 variables that might have had the potential to contribute to coal bumps (tables 1-3). In the second step, a "damage rating" was established as the dependent variable against which all other variables were tested to determine which of these variables were effective in deducing bump potential. The damage rating is a measure of the severity of coal bumps and was based on observations of physical damage to face equipment and/or injury to personnel, as well as observations by other researchers, as cited in the literature.

Coal bumps can lead to serious injuries to underground mine workers.

Table 1.—Statistical summary of geologic variables

Variable	Mean	Standard deviation	Range	No. of cases
Joint sets	1.4	0.6	1-3	25
Cleat sets	1.8	0.4	1-2	25
In-seam partings	1	0.9	0-3	21
Joint spacing, m	6.7	5.5	1.5-15	24
Rock Quality Designation	77	18	50-100	15
Depth, m	500	134	275-820	25
Roof beam thickness, m	4.3	3.3	1.5-12.2	25
Young's modulus, coal, MPa	3,310	830	2,410-4,620	25
Young's modulus, roof and floor, MPa	20,700	6,900	6,900-33,000	25
Uniaxial strength, MPa	22	5.2	13.8-32	25
Uniaxial strength, roof and floor, MPa	100	23.8	55-150	25
Maximum horizontal stress, MPa	13	7.6	0.7-26	25
Interacting seams	1.2	0.4	1-3	25
Local yield characteristics	0.8	—	0-2	25

Table 2.—Statistical summary of geometric variables

Variable	Mean	Standard deviation	Range	No. of cases
Pillar width, m	19	0	9-42	23
Pillar height, m	2.5	0.3	1.7-3	25
Entry span, m	5.8	0.3	5.5-6	25
Barrier pillar width, m	50	27	15-73	6
Face width, m	167	40	61-244	25
Mining method	1.2	0.4	1-2	25
Stress gradient	0.9	0.6	0-2	25

Table 3.—Statistical summary of geomechanical variables

Variable	Mean	Standard deviation	Range	No. of cases
Pillar factor of safety	0.8	0.3	0.5-1.4	23
Face factor of safety	0.9	0.2	0.6-1.5	22
Energy (M_1)	3.0	0.5	2-4	22
Damage	1.4	1.0	0-3	25

Damage levels were assigned a ranking for each case history. Next, a preliminary bivariate correlation procedure was performed among all variables. Based on these results, the number of variables was reduced by combining some variables into new ones and eliminating those that were intrinsically interrelated. In addition, the cause-and-effect structure in the data could be identified, thus helping to tailor the procedures for multiple regression analysis using forward stepwise inclusion of variables. The new variables were—

Pqratio Ratio of maximum principal horizontal stress (P) to minimum stress (Q).

Strenrc Ratio of uniaxial compressive strength of the roof to the coal.

Jointrf Joint spacing times roof beam thickness divided by mining height.

Gradyield Ratio of roof and floor yield characteristics to stress gradient.

Panelwd Ratio of panel width to depth.

Youngrc Ratio of Young's modulus of the roof to coal seam.

Bivariate correlation coefficients were calculated between the damage variable and the selected geologic and geometric variables. Energy (M_1), face factor of safety, stress gradient, pillar factor of safety, joint spacing, and uniaxial compressive strength of roof to coal were the most significant. Other variables were poorly correlated with damage, including the ratio of P to Q, pillar width, and Young's modulus of roof to coal seam.

Performing Multilinear Regression and Determining Influencing Factors

In the final step of the analysis, a multilinear regression procedure was used, which involved entering the independent variables one at a time as established in the previous step into an equation using a forward selection methodology. In this method, the variable having the largest correlation with the dependant variable is entered into the equation. If a variable fails to meet entry requirements, it is not included in the equation. If it meets the criteria, the second variable with the highest partial correlation will be selected and tested for entering into the equation. This procedure is very desirable when there is a cause-and-effect structure among the variables. An example of the cause-and-effect relationship is shown when a greater depth reduces pillar factor of safety, contributes to an accumulation of energy, and ultimately results in greater damage. Using the above procedures, any hidden relationship between depth and pillar factor of safety, energy, and damage can be evaluated and taken into account during each step of the analysis.

Several geomechanical variables (table 3) were initially used as dependent variables. The multiple correlation coefficient (R), which is a measure of goodness of fit, for the last step was 0.87. The assumptions of linear regression analysis were tested and found to be valid by an analysis of variance, F-statistics, and a plot of standardized residuals. Residual plots did not indicate the need for inclusion of nonlinear terms because there was no special pattern in the residuals.

Conclusions

Based on an examination of standardized regression coefficients, the following variables best explain variations in damage and thus statistically have the most significant influence on coal bumps.

- *Energy.* This variable includes the effects of the mechanical properties of the roof and coal, depth, stress field, and joint density, and therefore directly relates to damage.
- *Mining method.* Mining method has a bearing on coal bump potential. The room-and-pillar method is associated with a higher degree of damage than longwall mining.
- *Pillar factor of safety.* Gate pillar geometry contributes directly to the severity of damage.
- *Stress gradient and yield characteristics.* Mining toward areas of high stress creates a potential for coal bumps, whereas localized yielding roof and floor conditions encourage gradual failure, which reduces the amount of damage.

By combining the strength of computational and statistical methods, the research team is making significant progress toward predicting coal bump potential and building confidence intervals. Because the method relies on an extensive amount of geotechnical data, it is a practical approach to understanding factors that influence coal bump potential and will be helpful to mine planners in selecting relevant variables for assessing bump-prone conditions. This in turn will result in safer designs for coal mines.

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NIOSH

NIOSH Wins 1997 R&D 100 Award

The National Institute for Occupational Safety and Health (NIOSH) recently won a prestigious R&D 100 Award that recognizes the 100 most technologically significant new products of the year as judged by *R&D Magazine*. NIOSH received the award for the Portable Modified Direct Method Apparatus, which measures the gas contents of coal and other rock samples in field or laboratory settings. The device, developed by NIOSH's Pittsburgh Research Laboratory, is easy to use, rugged, compact, lightweight, reliable, and relatively inexpensive, yet provides more accurate gas content data than current instruments. Through these advances, this invention can reduce the risk of mine explosions, thereby improving safety for underground mine workers.

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